I. C. 6741 AUGUST, 1933

UNITED STATES BUREAU OF MINES
SCOTT TURNER, DIRECTOR

INFORMATION CIRCULAR

MINING METHODS AND COSTS AT THE McINTYRE PORCUPINE MINES, LTD., SCHUMACHER, ONTARIO

RECEIVED
[illegible]
O. S. U. LIBRARY

BY

H. G. SKAVLEM

I.C.6741,
August, 1933.

INFORMATION CIRCULAR

UNITED STATES BUREAU OF MINES

MINING METHODS AND COSTS AT THE MCINTYRE PORCUPINE MINES, LTD., SCHUMACHER, ONTARIO[1]

By H. G. Skavlem[2]

INTRODUCTION

This paper is prepared for the United States Bureau of Mines and is descriptive of methods, practice, and costs at the McIntyre Porcupine Mines, Ltd., Schumacher, Ontario.

The property is located in the Pearl Lake section of the Porcupine mining district. It is reached by a branch line of the Temiskaming & Northern Ontario Railway from Porquois Junction. There is also a good motor road into the district. The Hollinger Consolidated Gold Mines and the Coniaurum Mines, Ltd., are adjacent properties.

Power for both the mining and the pulp and paper industries in the district is supplied by an extensive development of the water-power resources.

Weather conditions are favorable for mining operations.

ACKNOWLEDGMENT

The author thanks the members of the operating and engineering organization of the mine, and especially R. J. Ennis, the general manager, for suggestions and criticisms concerning this paper.

HISTORY

The McIntyre Porcupine Mines, Ltd., the property of which was staked during the Porcupine rush of 1909, ranks second among the producers of the Porcupine district.

In 1914 the company owned four claims comprising a total of 145 acres. Since that time the property has been increased to 681 acres by the acquisition of the Pearl Lake Gold Mines, Ltd., Jupiter Gold Mines, Ltd., Plenaurum Gold Mines, Ltd., Platt Veteran Claims, and 52 acres under the eastern part of Gillies Lake, - all adjacent properties. The holding is in a continuous block and extends for 1 3/8 miles along the strike of the zone in which are found the principal producing veins of this section. Figure 1 is a sketch of property showing porphyry outcrops, location of shafts, and the old and new mill sites.

In March 1910 a 10-stamp amalgamation and concentration mill was erected. With further development a pronounced change in the percentage of sulphide ore was indicated and in 1913 a 150-ton, all-sliming-cyanidation plant was built. From time to time additions and alterations were made to this mill until its capacity was increased to 1,600 tons per day. The mill operated until November 15, 1931, but is now completely dismantled.

On May 27, 1931, the new mill of 2,000 tons capacity was started. On November 15 of the same year its rated capacity was attained.

---

[1] The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U.S. Bureau of Mines Information Circular 6741."

[2] One of the consulting engineers, U.S. Bureau of Mines, and chief engineer, McIntyre Porcupine Mines, Ltd.

The new mill and plant are of fireproof construction.

Present Milling Practice

The present milling practice differs from that of the original plant in that the gold and gold-bearing minerals are concentrated by flotation and only the concentrates are reground and cyanided.

The ore is first crushed in a 36- by 48-inch jaw crusher placed at the bottom of the ore pass at the 3,875 level of the mine. From a storage pocket below the crusher the ore passes to measuring boxes and from them into the 6-ton skips. It is then hoisted to a storage bin at the shaft head and fed from there to a No. 7 cone crusher by a pan conveyor. The product from the cone crusher is conveyed to a surge bin. At this step the ore is weighed by a Merrick weightometer. From the surge bin it is fed to six 6- by 4-foot Hum-mer screens. The undersize is 3/16 inch and is conveyed to the 4,000-ton storage bin; the oversize passes to a set of 78- by 18-inch rolls. The product from these is returned to the surge bin and again passes over the Hum-mer screens.

Secondary crushing is done by five tube mills in closed circuit with Dorr duplex classifiers. The tube-mill product, which is 90 percent minus 65 mesh, is pumped to the primary flotation cells and from them to secondary flotation cells. Native gold and the gold-bearing sulphides are concentrated by means of chemical flotation reagents. The ratio of concentration is about 8 or 10 to 1. Flotation tails go directly to waste.

Flotation concentrates are dewatered and washed and are ground in two tube mills in closed circuit with Dorr duplex classifiers. Cyanide is added at this point.

Combined loss from the concentrratess and flotation tails from an $8 to $9 ore is about 35 cents per ton; this loss will be reduced.

PRODUCTION AND DEVELOPMENT

Production at the property began in 1912 from veins in a small area of basic schist situated between the porphyry and the south boundary at the west end of the property and adjacent to the Acme claims of the Hollinger Consolidated Gold Mines, Ltd.

Development is by vertical shafts. Seven of the 16 shafts sunk to various depths on the property have been of major importance. The trend of development is illustrated by the depth of successive operating shafts:

Shaft 4, south and west end of property, reaches a depth of 820 feet.

Shaft 5, north of shaft 4 and 340 feet from the west boundary, reaches a depth of 2,388 feet.

Shaft 6, 825 feet northeast of shaft 5, is 3,015 feet deep.

Shaft 11, 2,000 feet east of shaft 6, is 4,133 feet deep.

Shaft 11 is the latest major development. It was started in October 1924 and includes a shaft house and hoisting equipment. Sinking was resumed February 2, 1925, and the shaft reached its objective of 4,133 feet on March 5, 1927.

Throughout the history of the mine the policy has been one of aggressive development. Beginning with a relatively small surface showing, the development to March 31, 1932, was as follows:

	Feet
Drifting	184,266
Crosscutting	86,819
Raises	19,112
Shafts	12,620
Diamond drilling	317,471

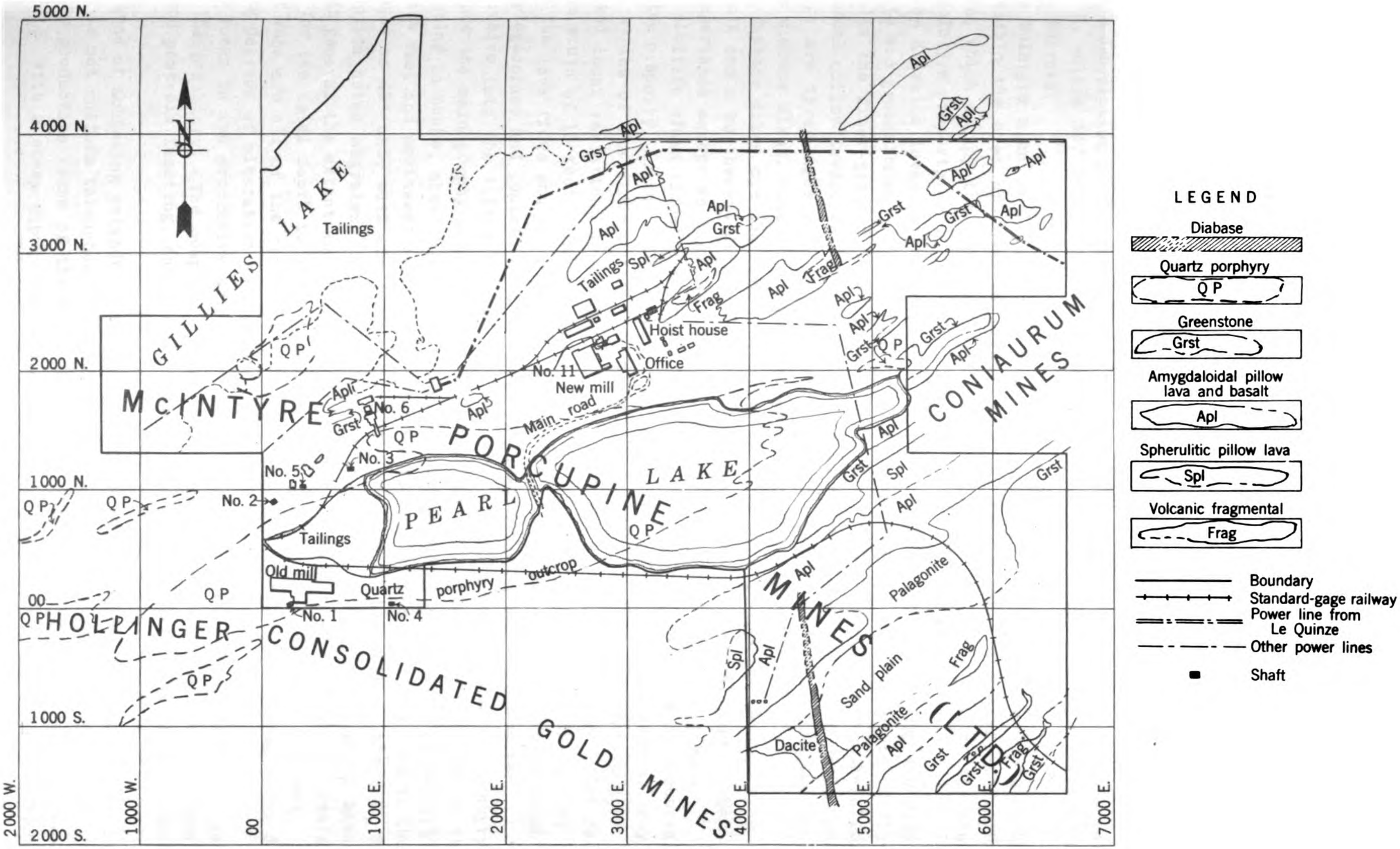


Figure 1.—Sketch of property showing porphyry outcrops, location of shafts and old and new mill sites.

The total production to March 31, 1932, was 6,015,718 tons, $8.95 average assay value, $53,957,841 total value.

The ore reserves as estimated on March 31, 1932, were 2,562,563 tons, $7.75 average assay value, $19,859,863 total value.

GEOLOGY

For a description of the geology of the Porcupine mining district the reader is referred to part 2, volume 33, 1924, of the 33rd Annual Report of the Ontario Department of Mines, by A. G. Burrows.

The McIntyre mine is located on the north limb of the Porcupine syncline. It lies wholly within the area of Keewatin lavas and later intrusives. The area of Temiskaming sediments, which occupy the central trough of the Porcupine syncline, lies to the southeast of the McIntyre property.

These Keewatin lavas have been subdivided in considerable detail. They are classified as basalts and greenstones. Greenstones are the coarse-grained flows or parts of flows and the basalts the finer-grained variations. These are further divided into spherulitic and amygdaloidal pillow lavas, palagonitic pillow lava, and some characteristic dacite phases.

There are three ages of intrusive activity with entirely different types of rocks. They are diabase dikes, albitite dikes, and quartz porphyry masses.

The diabase dikes cut the lava flows, the porphyry masses, and the veins. They have a north-south and a northwest-southeast strike and have steep or vertical dips. They are of little importance except as they interfere with operating conditions underground.

The albitite dikes (locally known as "goose egg") are found throughout the central portion of the property, north of the porphyry and below the 2,125 level. In general they parallel the strike of the flows - that is, N. 60° to 70° E. - and dip at steep angles. They have marked local variations in strike. The longest has a strike length of 1,000 feet and a maximum width of 15 feet. These dikes are relatively coarse-grained, with chilled edges; they cut the lava flows and the porphyry but are cut by the veins. They consist essentially of acid plagioclase and contain numerous rounded fragments of granitic character.

Intrusive into the tilted and folded lavas are several stocks of quartz porphyry. At the surface the main porphyry mass underlies Pearl Lake and extends to the west of it. It is lenticular in shape, about 6,000 feet long, and 1,500 feet wide in its widest part.

To the west and northwest of it there are several smaller masses. They rake to the east at an angle of 45° and, with one exception, the direction of the rake is about N. 84° E. The exception noted migrates to the north and joins one of the other porphyry masses at depth. In general the extent and configuration of the principal porphyry masses remain much the same for the total depth to which they have been delimited - that is, 3,875 feet. These masses of rock are all of the same type, the differences in the hand specimens being due to to varying degrees of alteration and schistosity.

The rocks in the productive area have been subjected to compressive forces at several times in their history. The complex folding, the fractures now occupied by the veins, the prevein and postvein faulting, and the broad belt of schist are evidences of these tectonic forces.

The zone of schisting extends in a northeast and southwest direction for several miles. Schisting is not uniform throughout this length, but in it over a width of 1,600 feet occur most of the productive veins of the Pearl Lake area. The strike of the schist is about N. 65° to 80° E. with a steep dip to the south, except in areas where it is changed by local folding.

The Pearl Lake porphyry mass is centrally located with regard to the Pearl Lake area and is within the zone of schisting.

To the north of the Pearl Lake porphyry the lava flows face north and are nearly vertical. To the south of the Pearl Lake porphyry they face south and dip to the south. This indicates a minor fold in this area on the north limb of the Porcupine syncline. The indications are that the Pearl Lake porphyry mass was intrusive along the axial plane of the minor fold. The elliptical shape of the mass and the fact that apophyses from it tend to parallel the flow structure are further evidence of this prior structural control. Also at the western end of the minor fold they are truncated by the porphyry intrusives. Figure 2 is a generalized structure section of the Pearl Lake area showing the folding and the intrusive porphyry.

The author believes the hypothesis to be reasonable that the lavas were folded and a regional pattern of stress control was in existence before the porphyry intrusion.

The later stresses then took place which were, in effect, a renewal of the regional stresses complicated by the forcs set up by the cooling porphyry masses and by the alteration effects of the porphyries on the intruded lavas. During this period of activity the physical character of the lava flows and their attitude - that is, folding and position with regard to the intruded porphyry masses - had a marked influence on the degree of fracturing of the schisted and unschisted areas both for the lavas and for the porphyry.

There are several planes of fracturing throughout the schisted zone. They are identified on the McIntyre by the several vein systems. These fractures cut across the schisting at a small angle to the left, and it is in them that the ore is found when the vein occupying the fracture is in a favorable structural relationship, usually of the Pearl Lake porphyry and the lavas. Development work on veins in the lavas and diamond-drill results in the intervening porphyry indicate the continuity of these planes of fracture or major breaks throughout the length of the McIntyre property, regardless of the type of rock.

Development has proved that vein material occurs in these major breaks in all the rock types or their derivatives in this area, but orebodies have only been found in the lava-schists or at the lava-schist porphyry contacts. To date no major orebody has been developed in the porphyry mass. Ore is found passing through porphyry tongues and it has been mined for some distance into the main porphyry mass from some of the higher-grade contact orebodies. Figure 3 is a plan showing generalized structural factors and Figure 4 is a cross section at shaft 11.

There are at least six major breaks within the zone of schisting on the McIntyre property. They have been determined by underground development, principally in the lava schists, but diamond drilling and some crosscutting in the main porphyry mass have indicated their continuity through this formation.

As development has proved that the most favorable ground for ore is along these breaks in the lava schists and lava-porphyry contact schists, it follows that any ground in which such structural conditions possibly exist must be explored and developed.

At the McIntyre there are two such general areas: (a) The ground to the south and west of the main porphyry mass or on the underlay side of the porphyry; (b) the ground to north and east of the main porphyry mass and generally above the porphyry.

For area "a" the breaks which are in the porphyry in the upper horizon of the McIntyre will pass into lava schist at lower horizons and farther to the east for each successive break from north to south. For area "b" these conditions exist from the surface to the lowest level of the McIntyre. The development policy of the McIntyre has been to open up these areas.

To date the greatest amount of development has been in area "a". Development of this area has been from shafts 1, 2, 3, 4, 5, and 6, the latter being sunk to the 2,875 level.

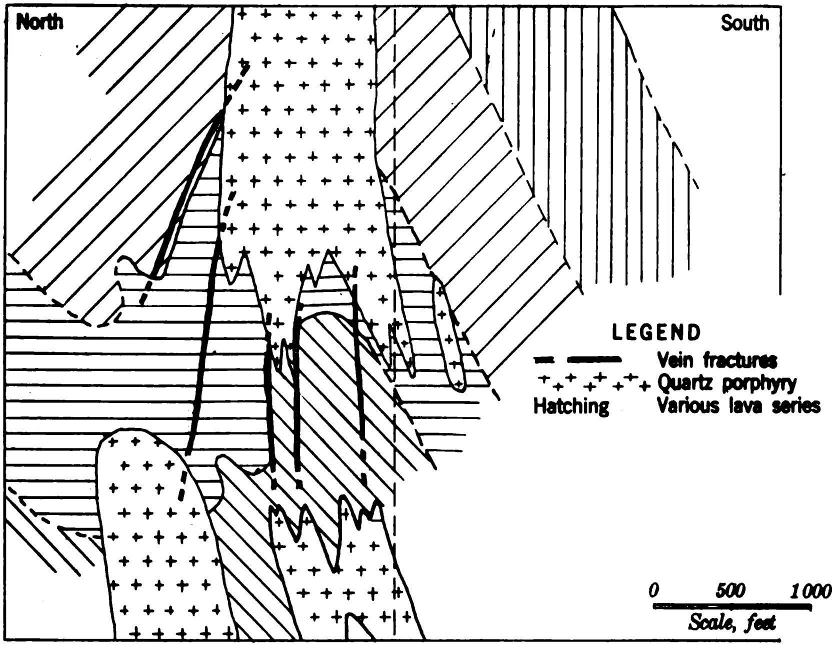


Figure 2.—Generalized structure section, Pearl Lake area, showing folding and intrusive porphyry.

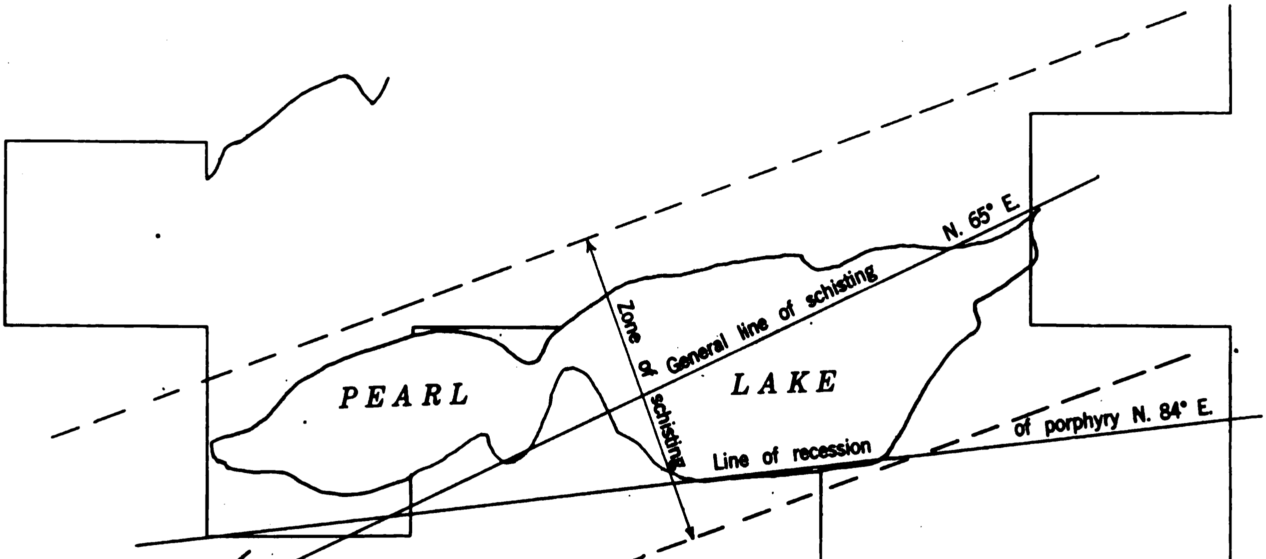


Figure 3.—Plan showing generalized structural factors.

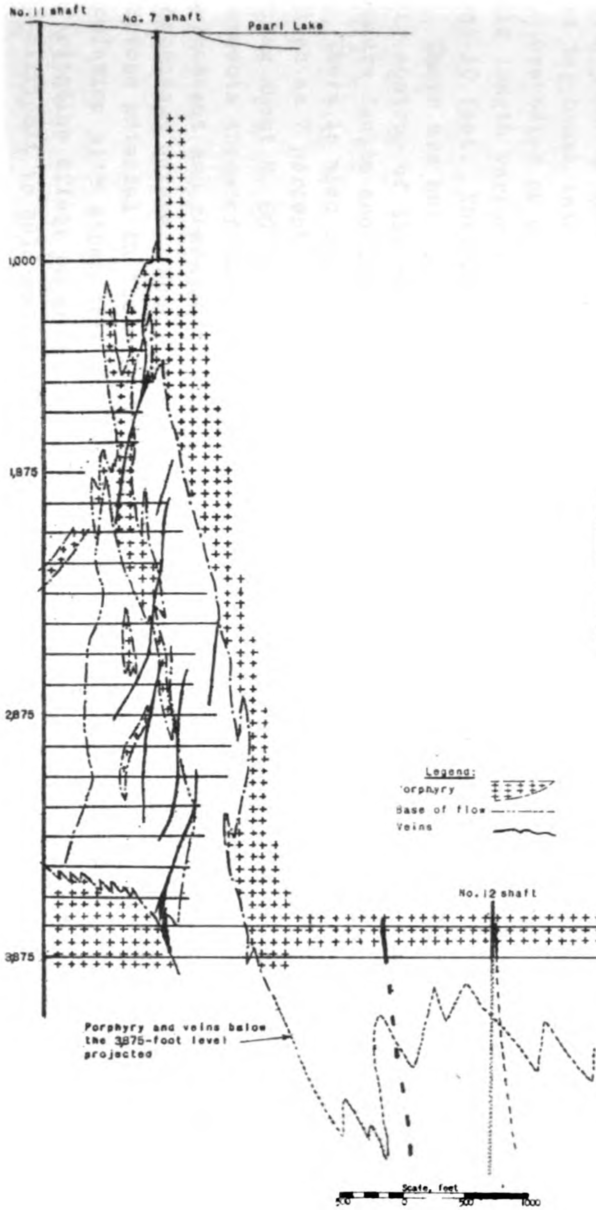


Figure 4.- Section through No. 11 shaft looking east

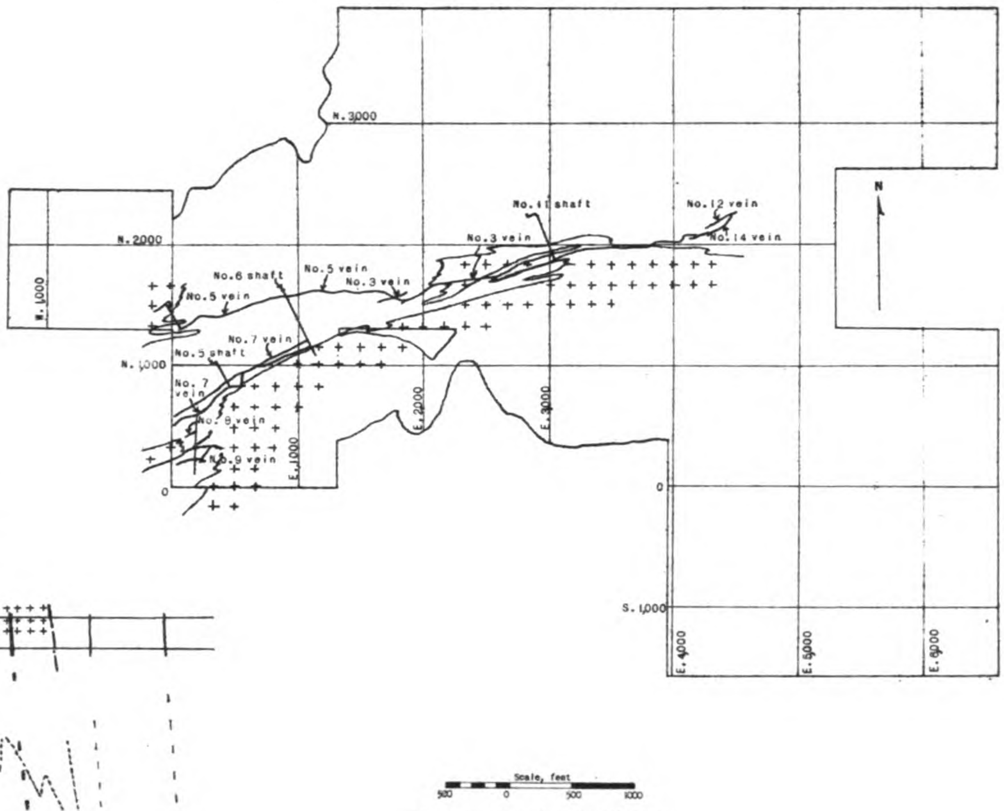


Figure 5.- Plan of 1500-foot level

Shaft 11 is so situated that area "a" is developed from it below the 2875 level and area "b" from all its levels. The first level from shaft 11 is at 1,000 feet so that development of area "b" above this horizon has been from shafts 7, 8, 9, and 10.

Development crosscuts to the southeast from shaft 11 on the 3750 and 3875 levels through the Pearl Lake porphyry and into the lava schists on its south side have proved several veins in the porphyry at these horizons and a parallel series of breaks to the south of the porphyry in the lava schists. At these horizons development of the farthest south break has proved ore for a length of several hundred feet. Diamond drilling to a depth of 4,400 feet has proved ore on this break to that depth.

The No. 12 internal shaft is now being sunk to develop the veins in the porphyry known to exist at the 3750 and 3875 levels at a horizon at which they will pass from the porphyry into the underlying schist and to continue development of the orebearing veins now developed on the 3750 and 3875 levels at lower horizons. This shaft is a four-compartment, rectangular shaft and will be equipped to handle 1,500 tons in 16 hours from a depth of 3,000 feet or to a maximum depth of 6,875 feet below the surface.

PHYSICAL CHARACTERISTICS OF ORE AND WALL ROCKS

The ore is of two types: Quartz veins with included schist fragments, and irregular replacement bodies. An orebody may be of both types.

As the orebodies occur in fractures which are common to all the rock types of the area, it follows that wall rocks are any of the types or their derivations. The following may be said in general concerning the principal alterations of the various rock types:

Basalts may show any type of alteration to any degree. Greenstones and dacites may show any type of alteration but to a much less marked degree than the basalts. Porphyries are not chloritized.

Wall rocks of the lava schists or their derivatives usually stand well unless broken up by faults. Wall rocks of contact orebodies or bodies in porphyry are usually carbonated and sericitized; they tend to be slabby and are difficult to hold. Disseminated sulphide orebodies usually have irregular walls due to the fact that mineralization often extends away from the break into the walls along joint planes and fractures.

Orebodies as a rule are nearly vertical, but some have a dip to the north of about 60°. Their length varies up to 1,200 feet and their width up to 100 feet. The average width is about 10 feet. They appear to have about the same characteristics at all horizons.

There are both premineral and postmineral faults. The former are important because of their control of the mineralizing solutions in some areas. The latter are practically all pressure faults and their displacement is usually not great.

There is also a zone characterized by an amorphous-carbon development which may contain as high as 7 percent of carbon in local areas where minor cross faulting occurs. This zone strikes about N. 60° E. and dips to the south. It is important because of the fact that it intersects three of the principal vein systems. Where this occurs there may be considerable displacement and consequent dilution of the original vein with wall rock of a more or less carbonaceous nature. Such dilution is important because of the precipitating effect of carbonaceous material in the mill solutions. It has been determined that amorphous carbon in association with other than cubical sulphide combinations of iron or copper has a marked precipitating effect on gold cyanide solutions. At these intersections the vein walls are also difficult to hold and such places have usually been mined by square-sets or square-sets and fill. These methods prevent excessive dilution, allow a certain amount of sorting, and make for safer mining conditions. There is a considerable tonnage of this material remaining in vein 5. With the new milling process it will be possible to salvage this ore. Because of its precipitating properties it could not be handled by the previous method of ore treatment.

PROSPECTING AND EXPLORATION

Development and exploration work on the property of the McIntyre Porcupine Mines, Ltd., is controlled by geological factors.

Areas favorably located from known geological evidence are opened up by crosscuts and drifts with no preliminary diamond drilling, but doubtfull areas are first drilled and as much information as possible is obtained before starting development. Certain other areas are drilled with the idea that negative results will be conclusive.

As the orebodies are successive lenses in parallel breaks, the method of development is usually to start drifting on what appears to be the best orebody in the parallel series. This is developed to its limit, and should there still be a break or indications of a feeder for the next lens, it is followed. Should there be no indications to follow, the drift is usually advanced about 50 feet in waste and then drill holes are put out to cut the parallel breaks in favorable ground. Subsequent development proceeds according to the results obtained.

The orebodies are lenses which step to the left in plan; they also have the same attitude vertically and the result is that many lenses have extensions which terminate between levels. Short diamond-drill holes are used to locate the bottoms of such lenses.

Diamond-drill holes are sometimes drilled to obtain geological information. Three diamond drills are employed. Size "E" bits are used in the majority of holes. For holes over 2,000 feet deep a "B" bit is used for the first 1,000 feet and then is changed for the smaller "A" bit. At present bort is being extensively used in drilling, especially in hard ground.

SAMPLING AND ESTIMATION

Not all vein material is ore. As a rule, however, ore can be identified as such from its appearance, although not always. Extensive sampling is therefore necessary to direct both development and stoping operations.

All development faces are sampled after each round is blasted. The samples are usually taken while the muck from the previous round is still in the face; consequently, at the time of sampling only one third of the face is exposed. No sample is longer than 5 feet, and most face samples are sectionalized so as to indicate the distribution of values. The first sample is always taken on the north side of the face.

Development samples or special samples are tied with a blue cord. All samples are taken to the surface at 11:30 a.m. Upon arrival at the assay office those tied with a blue cord are sorted out and assayed. The results are available for the night shift. The assays are typed on small loose-leaf sheets, and each shift boss has a binder for them. The underground operating department in this way has all the available information by which to direct mining operations.

Crosscuts are sampled at 5-foot intervals on the east rib. Check samples are taken on the west rib. All crosscuts are sampled.

In stoping operations an endeavor is made to sample each breast as broken. This is not always possible but should a stope be of low grade or have irregular walls it is sampled with greater regularity than a known high-grade stope.

Box or muck samples are taken from all development faces and also of the muck drawn from stope chutes. Drift walls are sampled at 10-foot intervals by test holes. Sludge for each 2 feet of length is collected and assayed. Stopes with irregular walls are also test-holed. These holes are usually marked up by the shift bosses.

The information from this work is used to direct slashing (slabbing) operations. Where considerable slashing is done, each slash is sampled. For purposes of calculation, where much slashing has been done, the backs are resampled at 10-foot intervals.

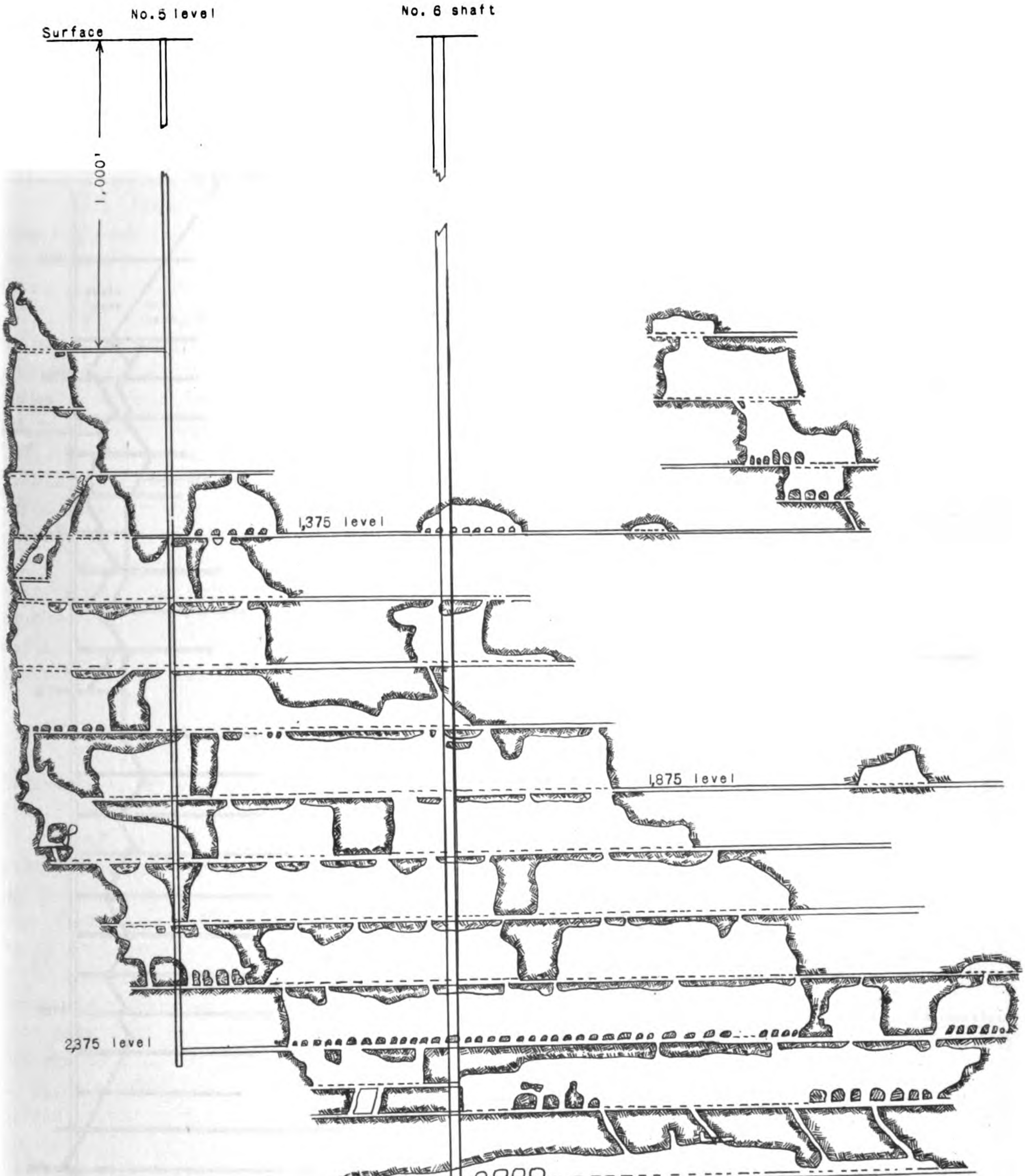


Figure 6.- Longitudinal elevation of No. 7 vein

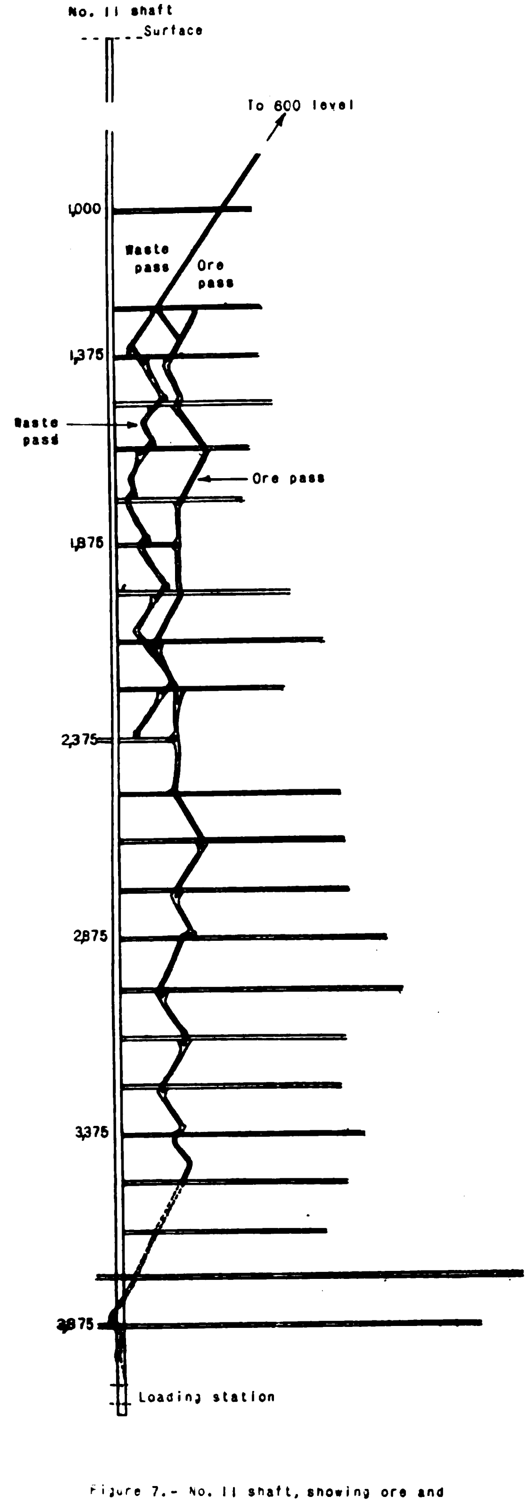


Figure 7.- No. 11 shaft, showing ore and waste passes

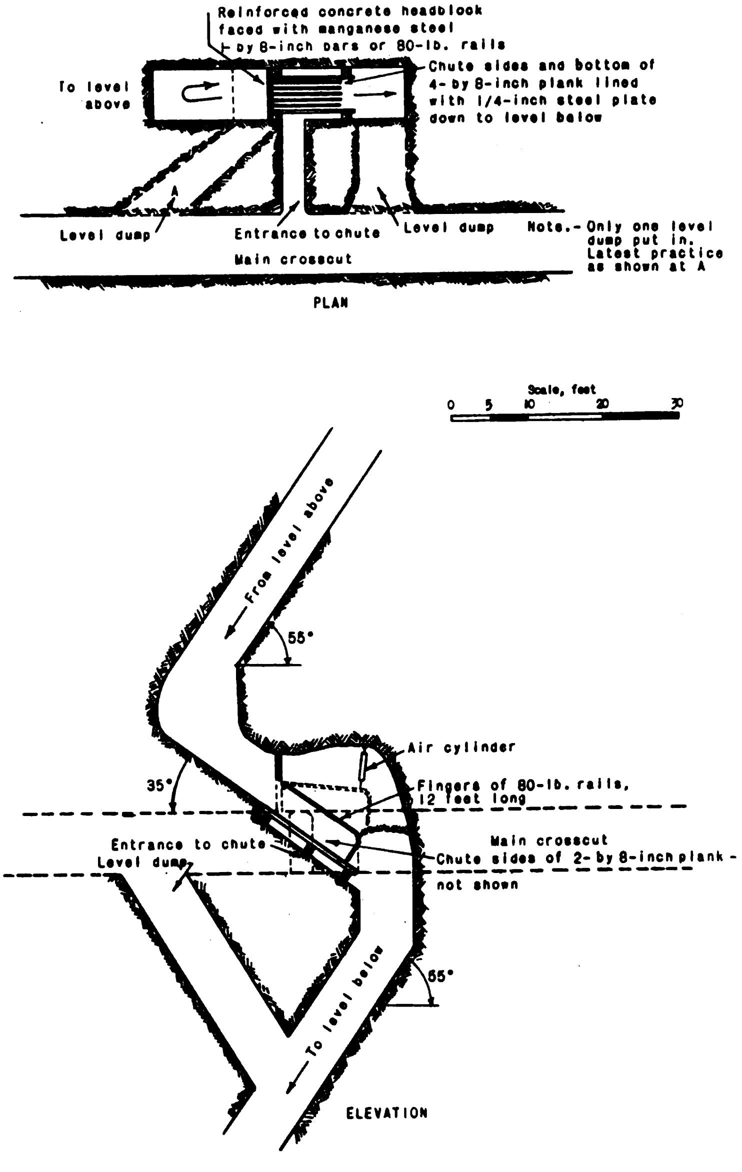

Reinforced concrete headblock faced with manganese steel 1- by 8-inch bars or 80-lb. rails
To level above
Chute sides and bottom of 4- by 8-inch plank lined with 1/4-inch steel plate down to level below
A
Level dump
Entrance to chute
Level dump
Note.- Only one level dump put in. Latest practice as shown at A
Main crosscut
PLAN
Scale, feet
0
5
10
20
30
From level above
55°
Air cylinder
Fingers of 80-lb. rails, 12 feet long
35°
Main crosscut
Entrance to chute
Level dump
Chute sides of 2- by 8-inch plank - not shown
55°
To level below
ELEVATION

Figure 8.- Standard air-operated control chute in ore pass.

In diamond drilling, a sample of sludge for each 10 feet of drilling is assayed. No attempt is made to collect all the sludge. The results of the sludge assays are obtained before examining the core, which is sectionalized for assay according to its appearance and sludge-assay results. Should the core be vein material but give no sludge values, it is sectionalized and assayed. A 5-foot section of schist on either side of vein material is also assayed.

A sample representative of each 5 feet of core, or of shorter intervals where a change in the nature of the core occurs, is kept for record.

All development samples are plotted on mine plans drawn on cross-section tracing linen to a scale of 1 inch to 10 feet. Before starting to take down backs in stoping operations, a blueprint of each drift with assay values is sent to the underground operating department.

For ore-reserve estimation the usual length, width, and value calculations are made.

Longitudinal sections of each vein are made on a scale of 1 inch to 60 feet. The vein is blocked out by coordinates 100 feet apart on an east-west plane. Levels are 125 feet apart. An effort is made to have a raise through between levels for each 300 feet of stope as soon as is practical.

Where erratic high assays occur they are cut to $50; where they are consistently high they are not reduced.

The known geological factors which affect the results are considered in addition to the mathematical calculations for tonnage and grade for each block.

Each stope is measured up every two weeks and the tonnage broken is calculated; these figures, together with the average assay of stope samples for that tonnage and the record of tonnage drawn from that stope with the corresponding muck assays, give an open inventory of the broken ore remaining in the stope. Erratic high box assays are arbitrarily reduced to $50.

No extensions of ore longer than 62.5 feet are figured above or below ore developed in a drift and no extensions greater than 50 feet beyond the faces are figured for faces in ore at the time of the reserve calculations.

From all of the foregoing factors a final estimate of ore reserves is made for each block in compiling the annual report.

Mine plans are drawn on a scale of 30 feet to 1 inch. One plan is made for each original claim. No plan has more than five levels plotted on it.

A general office plan on a scale of 100 feet to 1 inch is used for recording general information.

METHODS OF MINING AND DEVELOPMENT

The mine has been developed by vertical shafts. Levels are spaced at 100-foot intervals to the 1000 level and at 125-foot intervals below this horizon. Figure 5 is a plan of the 1500 level and Figure 6 is a longitudinal elevation of vein 7.

At present all ore is trammed to ore passes. In the western end of the mine an ore pass roughly parallels shaft 6 from the 400 level to the 2875 level. The ore is transferred on this level to the shaft 11 ore pass.

The ore pass at shaft 11 is driven at an inclination of 50° to 55° and extends from the 600 level to the top of the ore pocket at the 3875 level.

A waste pass driven at 50° to 55° inclination parallels the ore pass and extends from the 1250 level to the 2375 level. A connection with a deflecting chute between the waste and ore passes at the 1250 level allows the use of the pass above this level for both ore and waste. Figure 7 is a sketch of shaft 11 showing the ore and waste passes and loading pocket.

There are control chutes in the ore pass at every fourth level. These are finger chutes and are usually placed below a change of direction in the pass. Figre 8 shows the standard air-operated control chute.

It is necessary to keep the ore as dry as possible. For that reason all water is drained away from the pass; consequently, considerable dust is raised when ore is running in the pass. To prevent this dust from entering the mine workings a special door has been devised for the ore-pass dumps on each level (see fig. 9). The frequency of their use determines whether these doors are hand operated or air operated.

At the 3875 level there is a control chute above the ore pocket. Formerly when operating at a capacity of 1,600 tons per day, the ore from this control chute was allowed to run out over an inclined grizzly to a flat sledging grizzly with 12- by 12-inch openings. Results showed that with the openings of this size about 1 percent of the tonnage required sledging. Feed from the pass to the grizzly was intermittent, and at times because of segregation of fines in the pass it hung up and required blasting.

With the increase in output to 2,000 tons per day the grizzly arrangement has been replaced by a 36- by 48-inch jaw crusher. Feed to the crusher is controlled by a chain feeder so that there are now no delays at the ore pass.

Sublevel development is extensive in some areas, even where the level interval is 125 feet. Short diamond-drill holes are used to determine the extension of an ore lens below the level. Should it extend more than 90 feet, a drift is driven on the level below and the ore is mined by means of box hole raises driven to the bottom of the lens and then connected in ore. Should it be less than 90 feet, a sublevel with timbered backs is used to mine the ore.

Manways are usually spaced 150 to 200 feet apart. Figure 10 shows manway and chute timbering and cribbing.

Raises are driven through to the level above as soon as practical. They are usually spaced about 300 feet apart and are driven without timber at an angle slightly under 50°.

Ore and waste passes are driven at an angle of 50° or more; they are timbered and blasted by electricity after the first 50 feet of advance.

SHAFTS

Shaft 11.- The excavation for shaft 11 was 17 by 24 feet. The short side is parallel to the schistosity of the country rock. Figure 11 is a sketch of the timbering and arrangement of shaft 11.

A summary of sinking operations from 58 feet in depth and after the headgear was installed follows:[3]

Number of rounds		400
Depth of sinking	feet	4,071
Holes per round	number	66.6
Drilling per round	feet	771.2
Steels used per round	number	113.5
Drilling	Total feet	303,886
Rock moved	tons	160,000
Water raised	do.	40,000
Timber placed	board feet	2,256,000
Men per shift	number	16.5

---

[3] Kee, H. A., Sinking Operations at McIntyre No. 11 Shaft, Porcupine District, Northern Ontario: Trans. Canadian Inst. Min. and Met., Bull. 192, 1928, 18 pp.

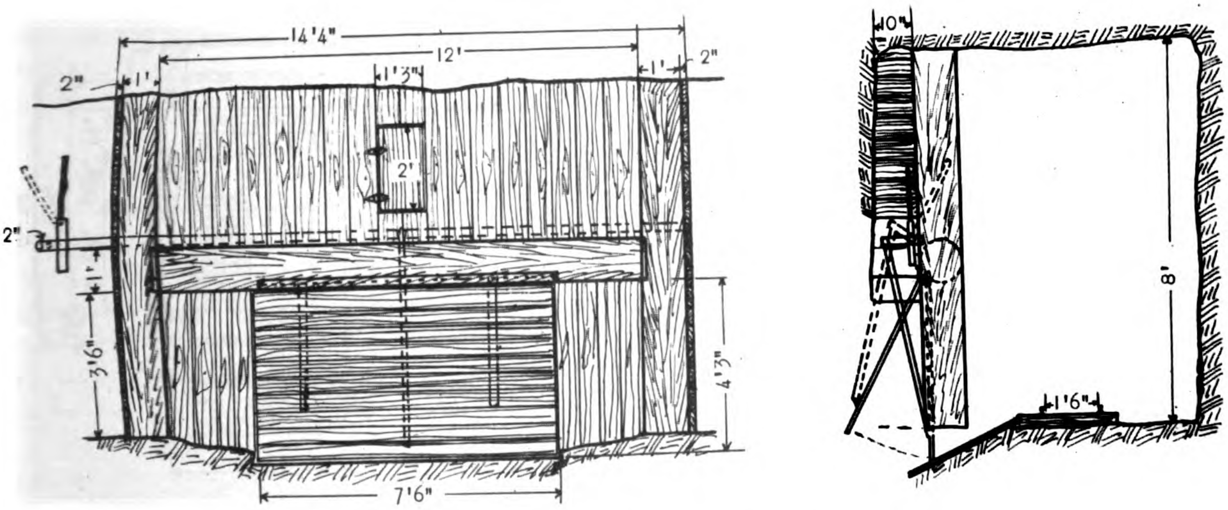


Figure 9.- Dust door for ore pass

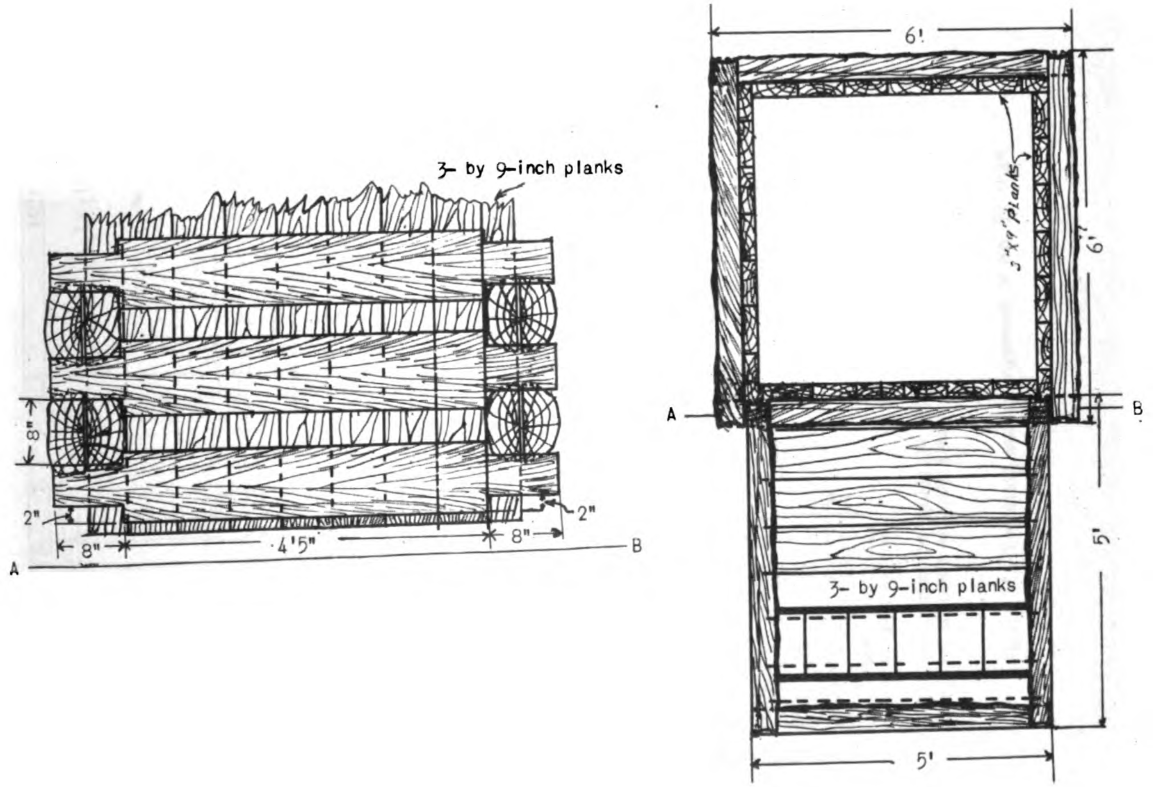


Figure 10.- Manway and chute timbering and cribbing

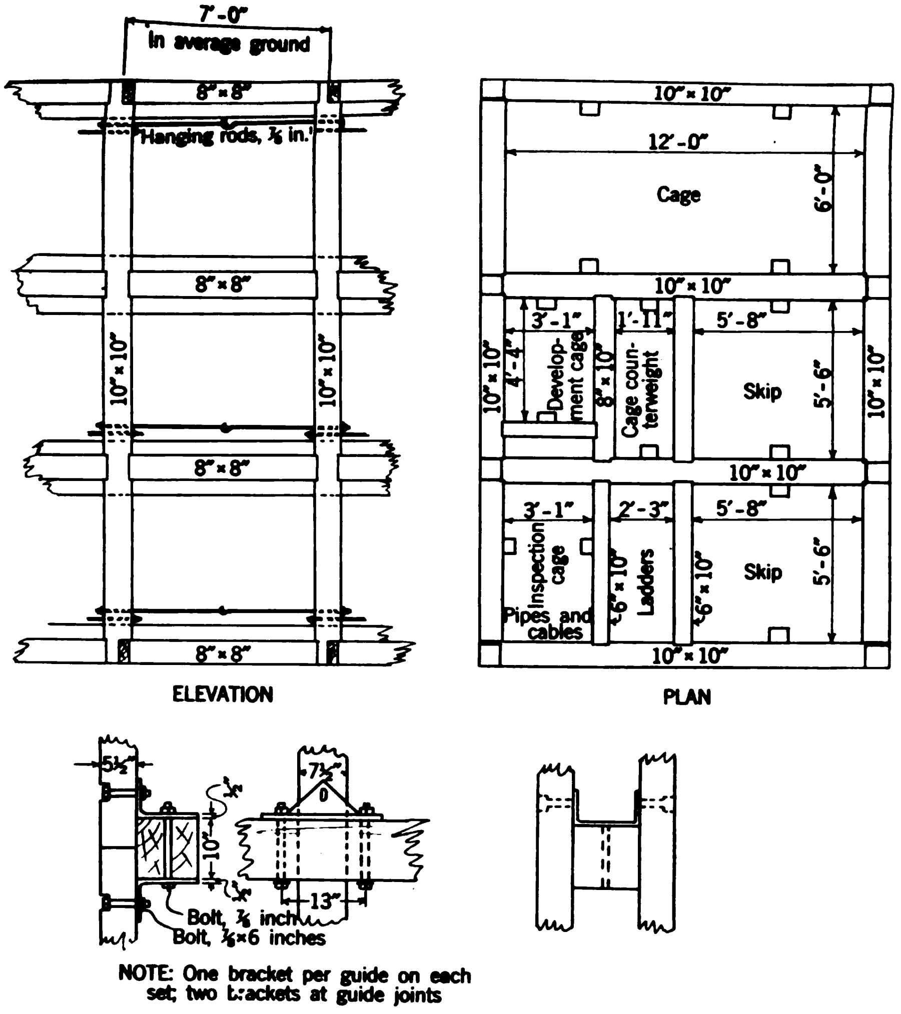


Figure 11.—Timbering and arrangement of No. 11 shaft.

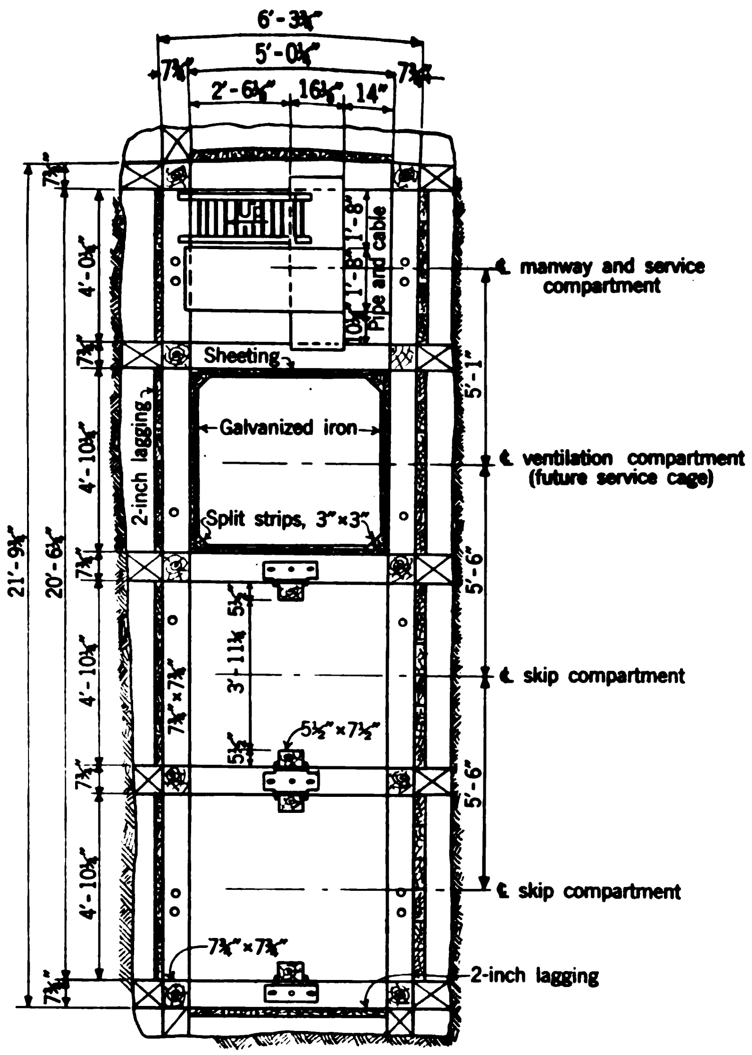


Figure 12.—Plan of No. 12 shaft.

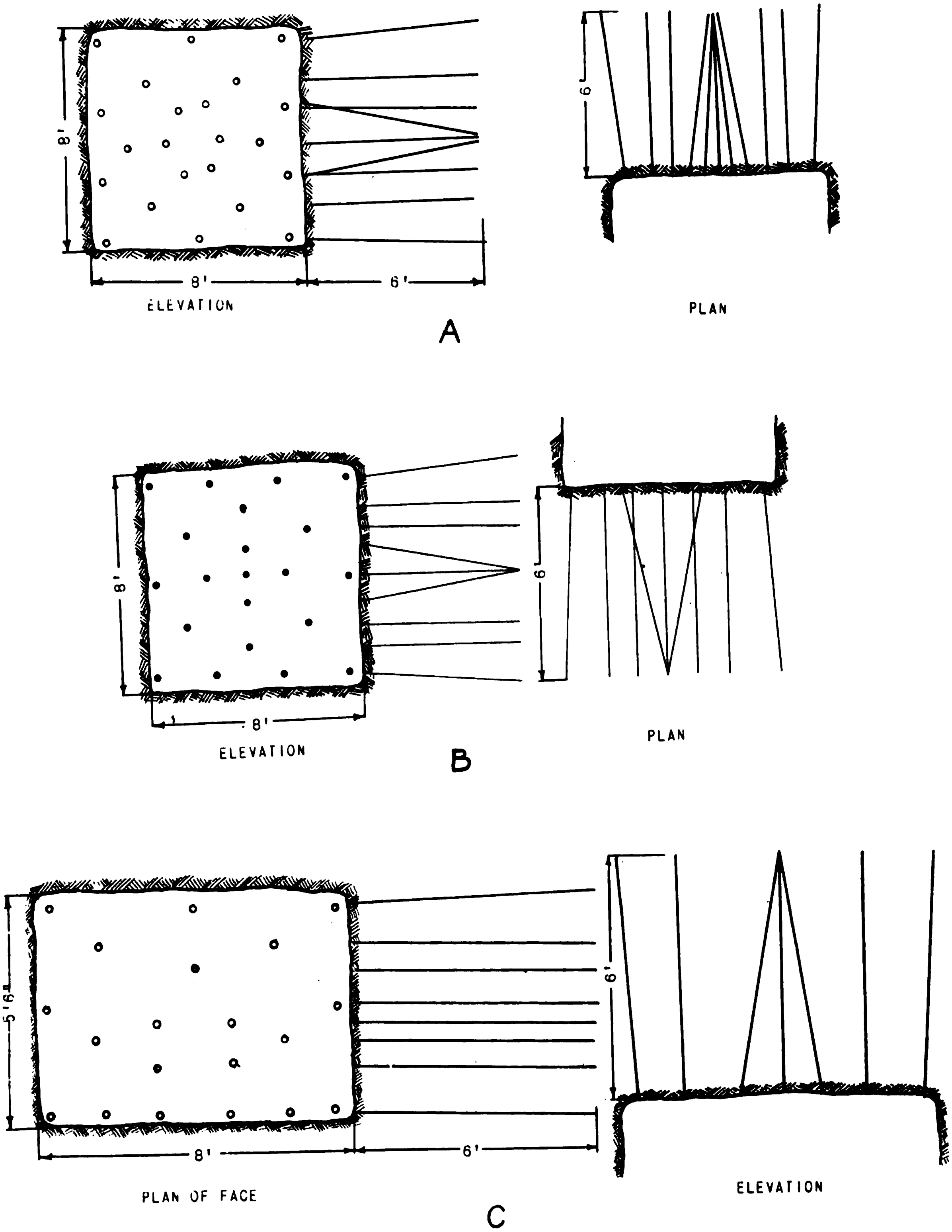


Figure 13.- Drill rounds: A, 22-hole drift round; B, 21-hole drift round; C, 20-hole raise round

Time cycle of operations:

	Percent
Drilling	9.0
Blasting and gas	11.3
Scaling	8.5
Mucking	49.2
Timbering	15.5
Hitch cutting	2.7
Delays	3.8

Total actual shaft time——700 days or 92 percent of total time.

Double and triple V-cut rounds were used. The blasting cycle was as follows:

Short cut-holes were blasted first and mucked out fairly clean. The long cut-holes and bench holes were then blasted. Light end holes to square up the shaft and also sump holes for the next round were drilled while mucking out the material from the second blast. These were then blasted and mucked out clean so that careful inspection of the bottom could be made before drilling the next round.

Electric blasting was done, wiring connections being in parallel series. The switch was in the mine captain's office at the surface, and all men were above ground before any blasting was done. Of the equipment used, the blasting shield, special crosshead arrangements to engage the bail of the bucket to prevent rotation while hoisting, special box for handling steel, and headers for air and water attachments for the drills and blow pipes are of particular interest.

Buckets held 1.6 tons and were designed with straight sides.

The shaft timber was Douglas fir, squared to four sides. Sets were spread on 5-, 6-, or 7-foot centers depending on the condition of the walls. The shaft was lagged outside with 2- or 3-inch plank and the space between lagging and walls was packed with split cedar logs. The timber was framed by machinery in a framing shed on the surface, and all sets were assembled and checked on the surface before being sent underground. Guides are of 5½- by 7½-inch yellow pine and are bolted to heavy steel brackets which in turn are bolted to the shaft timbers.

Timber was alined by six plumb lines having 6- to 8-pound plumb bobs. After alinement, sets were checked by steel-wire plumb lines having wing plumb bobs. Plumbing wire was on reels mounted on steel brackets with side and end adjustments. These were placed in the shaft corners to check the alinement from level to level.

Shaft 12.- Shaft 12 is an inside shaft, a plan of which is shown in figure 12. The hoisting equipment is on the 3750 level. Ore and waste pockets between the 3750 and 3875 level will provide hoisting flexibility. The headgear on the 3750 level is of all-steel construction. The rock excavation extends to a height of 94 feet above the 3750 level. All crosscuts, station raises, and ropeway raises have been covered with reinforced gunite.

Sinking to a depth of 1,500 feet below the 3875 level will be the first objective of this new development.

Drifts and Crosscuts.- Drifts are driven 8 by 8 feet in cross section. A pyramid cut with a round of 20 to 24 holes is generally used (fig. 13, A and B). The average round is just under 6 feet. The cut is blasted first and reblasted if necessary before blasting the square-up. After the initial blast, the men use respirators while completing the loading and blasting of the round.

A water spray is turned on immediately after blasting.

Machines with a piston diameter of 4 inches are used, but the tendency now is to employ machines with 3½-inch piston diameter.

The explosives used are 1¼-inch, 40-percent Polar forcite gelatin and 1¼-inch, 55-percent ammonia powder.

Mucking is done by hand. Mechanical loaders have been tried with some success.

Raises.- Raises are driven at an angle of approximately 48°. They are usually placed at intervals of 300 feet along the stope and are centrally located for the individual ore lens. They are started sufficiently wide for a standard chute and manway to be built at the level. Above this they are reduced to about 6 by 8 feet. Some shrinkage stopes are carried to a height of about 60 feet before raising through to the level above.

Ore-pass raises are driven at an angle of 50° plus, and are about 6 by 8 feet.

The practice in the past has been to drive all raises without timber.

Steel pins projecting 12 to 18 inches from the footwall of the raise and at the proper elevation, together with a chain ladder, provide access to the face. Staging is supported by sprags wedged from wall to wall, and lagging for sprags is used only once.

A round of from 18 to 22 holes is drilled, using a V-cut (fig. 13,C). The cut is blasted first and reblasted with the square-up, if necessary. Fuse with an average burning speed of about 40 seconds per foot is used. After a raise has reached a height of 50 feet, fuse in 10-foot lengths, before timing cuts are made, is used. Wet stopers are employed.

STOPING

The stoping methods used include shrinkage, cut-and-fill, square-set and fill, and square-set stoping.

Cut-and-fill Stoping.- As facilities are provided, the percentage of ore mined by the cut-and-fill method is constantly increasing. At present approximately 50 percent of the stoping is cut-and-fill.

The factors influencing selection of this method of stoping are:

Orebodies with irregular walls.
Less dilution by waste rock in sections with bad walls.
Waste may be sorted out.
Stabilization of workings in bad ground when they must remain open for some time.
Greater safety considering type of ground.
Manways are spaced about 200 to 250 feet apart.
Raises for back-fill and ventilation are 250 to 300 feet apart.

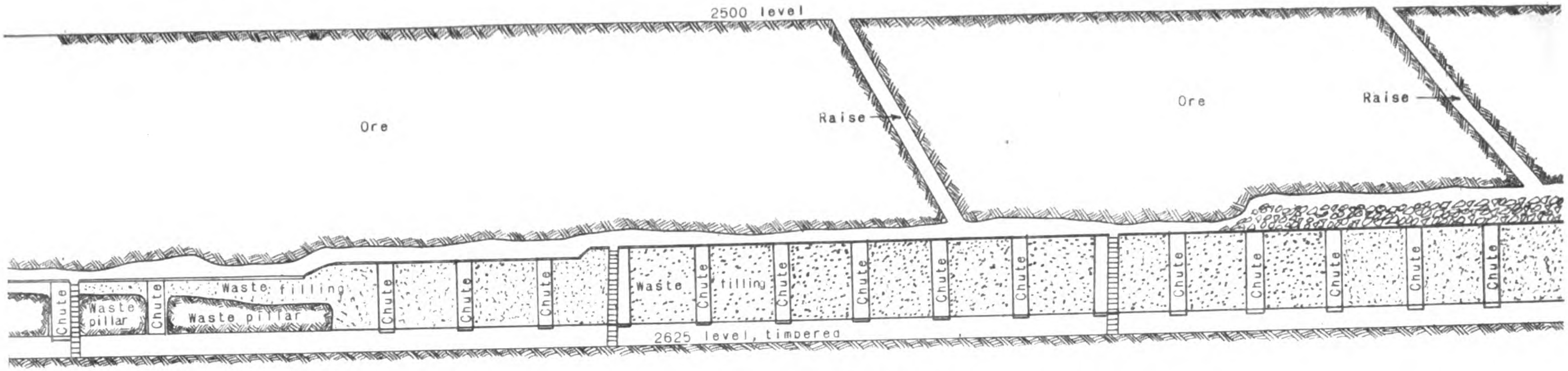
Breasts are about 8 to 10 feet in depth. All drilling is horizontal and the tonnage broken is about 9 tons per man-shift, exclusive of delivery. Slashing is regulated by test holes in the walls and is usually marked up by the shift boss. Waste is trammed from waste raises and is spread by hand. Filling operations and stoping operations are carried on simultaneously. About 1 pound of 1¼-inch Polar forcite gelatin is used for each ton broken. Figure 14 shows a cut-and-fill stope, No. 2522, in plan and elevation.

Square-Set and Fill Stoping.- The square-set and fill method is used only where special conditions such as faulted ground or ground under pressure are found. Where ground is particularly heavy, timber packs are built up from the fill to support the backs.

This method is especially applicable to stopes in which there is carbonaceous material, because of the better facilities afforded for sorting.

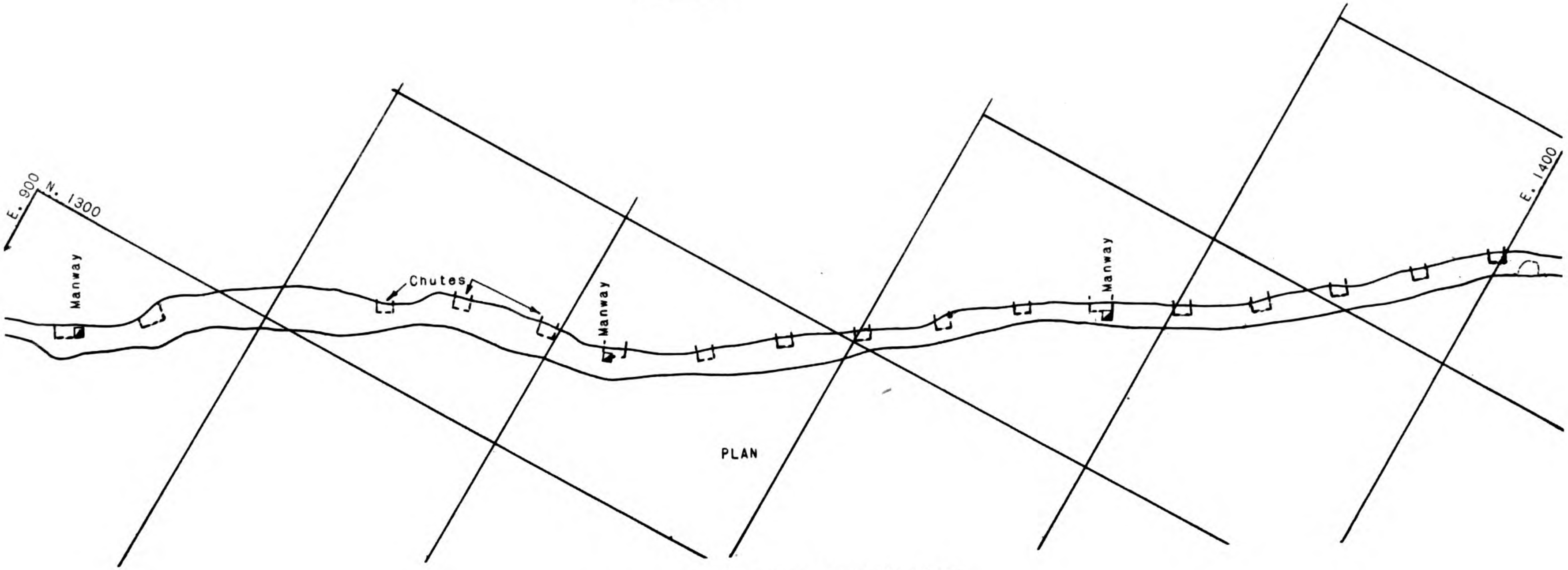
About 6 tons are broken per man-shift, exclusive of delivery, and the powder consumption is about ½ pound per ton broken. Figure 15 shows square-set and fill stoping and shrinkage stoping in stope 2028.

Waste for fill is obtained from development faces and from two shrinkage stopes in which waste rock is mined. Where waste is not available a system of small hanging-wall stopes to provide waste for each fill bench is now employed.



VERTICAL LONGITUDINAL SECTION

Scale, feet
0 10 20 30



PLAN

Figure 14.- No. 2522 cut-and-fill stope

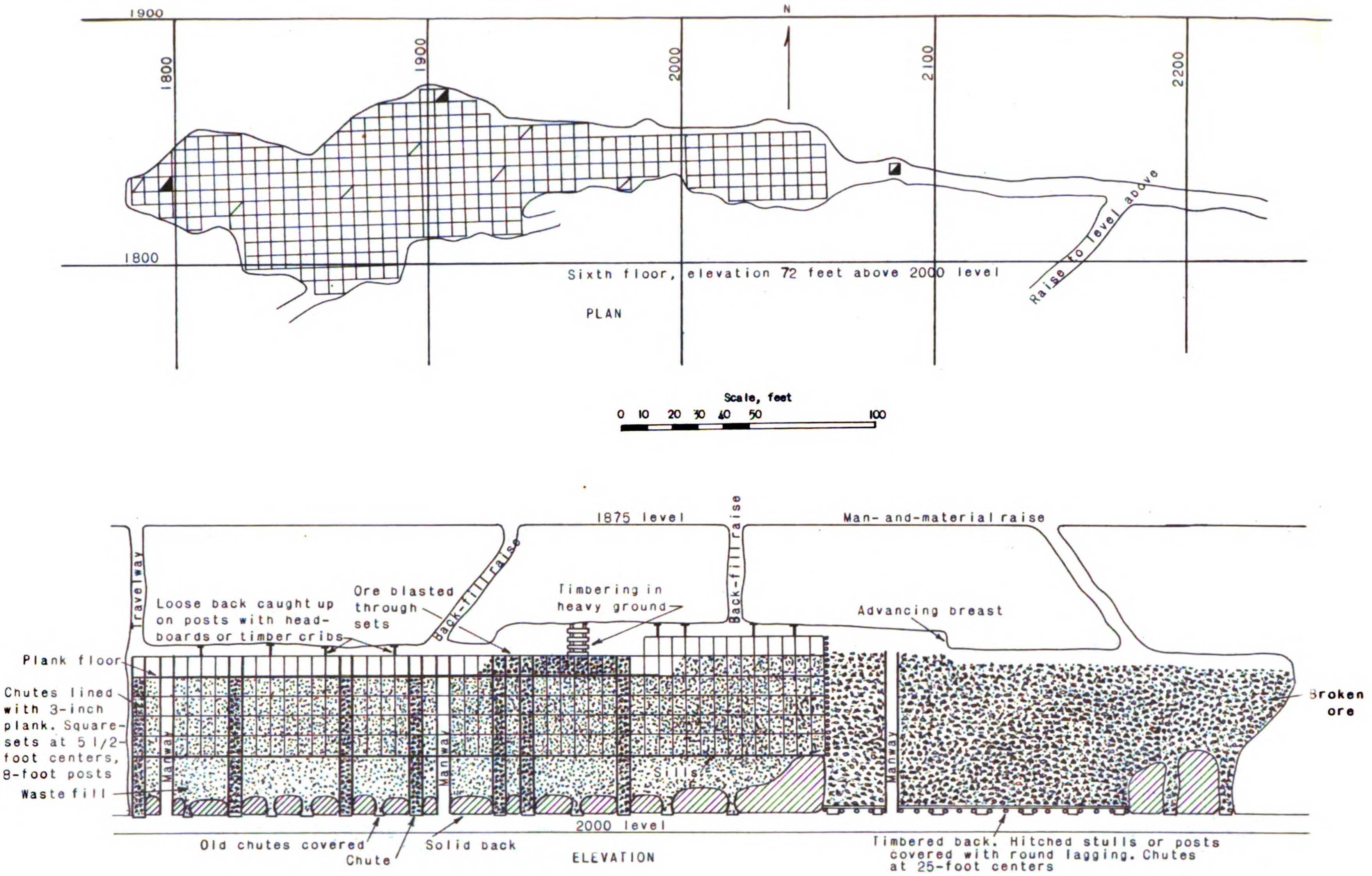

1900
N
1800
1900
2000
2100
2200
1800
Sixth floor, elevation 72 feet above 2000 level
Raise to level above
PLAN
Scale, feet
0 10 20 30 40 50 100
1875 level
Man-and-material raise
Travelway
Back-fill raise
Back-fill raise
Loose back caught up on posts with head-boards or timber cribs
Ore blasted through sets
Timbering in heavy ground
Advancing breast
Plank floor
Chutes lined with 3-inch plank. Square-sets at 5 1/2-foot centers, 8-foot posts
Waste fill
Manway
Manway
Manway
Sills
Broken ore
2000 level
Old chutes covered
Chute
Solid back
ELEVATION
Timbered back. Hitched stulls or posts covered with round lagging. Chutes at 25-foot centers

Figure 15.- No. 2023 stope, showing square-set and fill stoping (west end) and shrinkage stoping (east end).

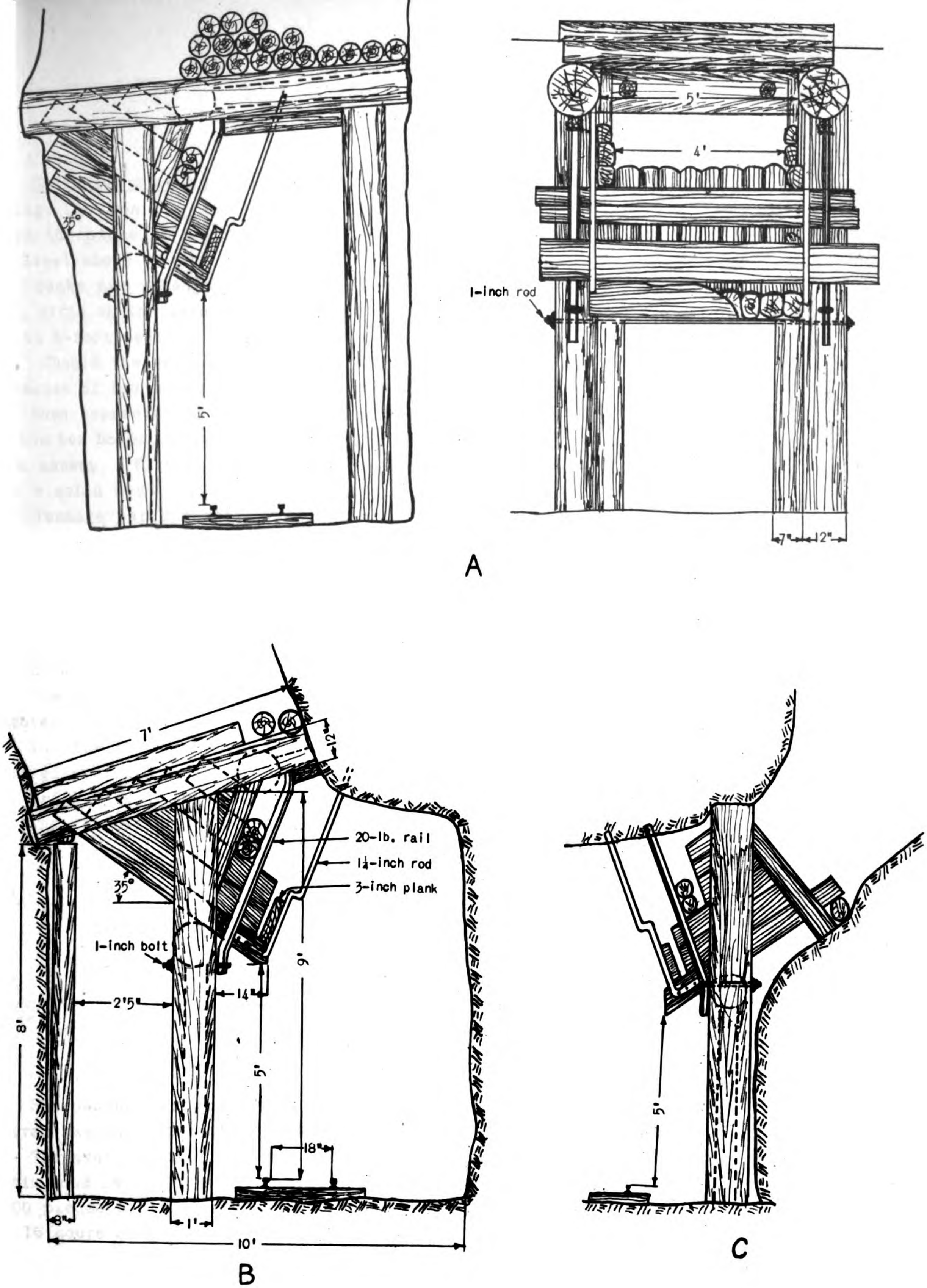


Figure 16.- Timbered chutes: A, Post and cap timbering; B, timbered back; C, solid back

Shrinkage Stoping.- Until recently shrinkage stoping has been the principal method of mining. Raises are sometimes put through from level to level before stoping starts, but often the stope is advanced to about 60 feet in height before the raise is put through to the level above.

Backs are taken down to a height of 18 to 20 feet after the vein has been slashed to full width on the level. The practice now is to timber the sill with posts and caps on about 4 - to 5-foot centers. Chutes are built 15 to 30 feet apart.

Should the vein be wide and of comparatively low grade, the backs are not shot down but a series of box holes are put up on about 25-foot centers for a height of about 30 feet and are then connected by sublevels. Regular shrinkage stoping then proceeds. At the bottom of the box holes, chutes are built and where necessary are widened out for the establishment of a manway. Figures 16,A, and 16,B, show timbered chutes and Figure 16,C, shows a chute with a solid back.

Tonnage per man-shift, exclusive of delivery, is about 20 tons and powder consumption about 1 pound per ton.

Pillars of low-grade ore are left for wall support and in some places timber packs are used.

At present the policy is to use this method in isolated stopes and stopes with probable regular and good walls, or in low-grade stopes of considerable width.

Breast stoping is used and holes are fanned out in wide stopes.

The machines employed are of several standard types and are usually of 3½-inch piston diameter.

In some of the longer shrinkage stopes, areas of local stress developed. Where this occurred, pillars were left.

Drill Steel.- One-inch, quarter-octagon, hollow steel is used with standard cross bits. The following lengths and gages are standard:

Feet	Bit size, inches
2.5 - 3.5	1-3/4
3.5 - 4.5	1-11/16
4.5 - 5.5	1-5/8
5.5 - 6.5	1-9/16
6.5 - 7.5	1-1/2
7.5 - 9.0	1-7/8
9.0 -13.0	1-3/8

Sharpening equipment consists of one 6-foot Donovan heating furnace and one 6-foot Donovan hardening furnace with magnetic control.

The average number of steels used per machine shift is 25 to 30, the average number of steels used per hole is 2.3, and the average distance drilled per steel is 2.91 feet. About 2,000 pieces of steel are sharpened per 24 hours. Three drill sharpeners are in operation for 16 hours each.

For the former mill capacity of 1,600 tons per day and a total yearly development of about 29,000 feet, a daily average of about 56 machine shifts was required.

The mean average air pressure was about 82 pounds.

The average tonnage produced per machine shift was about 35 tons (stoping).

The average footage drilled per machine shift was 75 feet.

The average tonnage broken per foot of drilling was 0.5 ton.

UNDERGROUND TRANSPORTATION

Tramming.- Grades are established at approximately one half of 1 percent in favor of the haul to shaft 11. Thirty-pound rails are standard. All curves have at least a 30-foot radius and all tracks are 18-inch gage.

Three types of cars are in use underground: 16-cubic foot, end-dump cars; 20- and 30-cubic foot, rocker-type cars; and 40-cubic foot, Granby-type cars. All cars are equipped with roller bearings.

The mine has the following locomotives for haulage purposes:

2 - 4-ton, two 20-hp. motor, trolley locomotives.
2 - Converter sets, 40-kw., d.c. generators.
4 - 2½-ton, 40-cell, worm-and-chain drive, storage-battery locomotives.
2 - 20-kw. charging sets, dynamo and motor.
2 - 1½-ton, single-motor, 3½-hp., storage-battery locomotives.
2 - Hertner vertical automatic generator sets.
1 - Mancha little trammer, 40-cell, storage-battery locomotive.
1 - Hertner vertical automatic charging set.

On levels other than those equipped with locomotives and connecting with shaft 11 and which have isolated working faces with long hauls, a horse is used for underground tramming and is transferred from level to level on the man-cage at shaft 11.

Tramming tonnage and cost ratios

		Tons ratio	Cost ratio
On tonnage basis	(Motor	44	1.6
	(Horse	10	1.4
	(Man	15	1.7
On basis of ton-miles..	(Motor	14	1.0
	(Horse	1½	1.1
	(Man	2	3.2

At present men and material are handled in shafts 5, 6, and 11.

Hoisting.- The hoisting equipment at shaft 11 is as follows:

Shaft 11 hoists

	Rock hoist	Man hoist
Weight of rock....................................pounds	12,000	Load 9,000
Weight of skip.................................... do.	8,000	Cage 16,000
Diameter of rope.................................inches	1½	1¾
Weight of rope per foot...................pounds (flattened strand)	4.2	5.7
Maximum winding speed.......................f.p.m.	3,000	2,000
Diameter and width of drums...............feet	12 by 8	12 by 8
R.p.m. at full speed....................................	79	52.5
Motor capacity.............................horsepower	2,450	1,400
Generator sets...............................kilowatts (Ward-Leonard)	2,000 (Equipped with 40-ton flywheel)	1,200 -

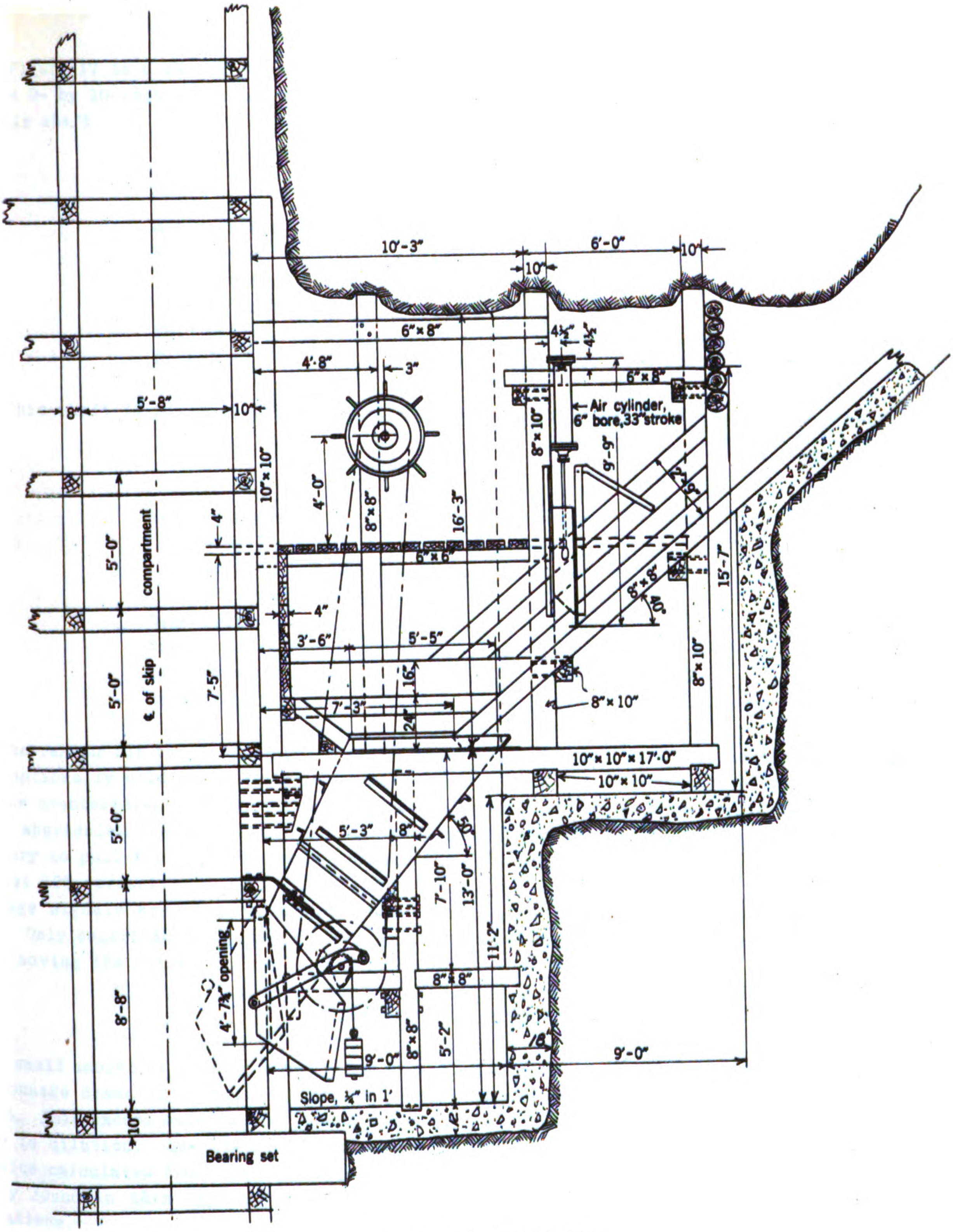


Figure 17.–Sketch of loading station.

Figure 17 is a sketch of the loading station.

A 9- by 10-inch air hoist to operate the auxiliary cage is installed on the 1250 level of this shaft.

Shaft 6 hoist

Motor capacity	horsepower	350
Rope speed	f.p.m.	1,800
(Double-drum hoist, handles two skips in balance)		
Skip weight	pounds	4,500
Rock load	do.	6,000

This shaft is now used for a service shaft and one skip has been replaced by a cage.

Shaft 5 hoist

Motor capacity	horsepower	125
(Handles 3,000-pound cage in balance with 3,750-pound reticulated counterbalance)		
Cage drum diameter	feet	6
Counterbalance drum diameter	do.	5
Rope speed	f.p.m.	750
Rope diameter	inch	1

The reason for the difference in diameter between cage and counterbalance drums is that in exceptionally cold weather there may be considerable frost near the top of the shaft. To pull the counterbalance through this section would cause excessive rope strain. Consequently, by shortening the travel of the counterbalance compared to that of the cage it is not necessary to pull the counterbalance through the full height of the shaft. The power factor is about 105 percent.

Cage signals are so arranged that cages may be signalled by buzzer or light from any level. Only cagers and skip tenders may signal the hoistman, and he must repeat all signals before moving the hoist.

PERCENTAGE OF EXTRACTION

A small amount of sorting takes place underground in cut-and-fill and square-set stopes.

Tonnage drawn from shrinkage stopes usually exceeds the calculated break by 10 to 15 percent. This excess material is usually lower in grade than the stope sampling indicates and is due to dilution. Quartz-type veins as a rule run about 10 percent higher in grade than the value calculated from the sampling. This is due to the fact that high-grade pockets are usually found in this type of ore deposit and because erratic high assays are shaded in calculations.

For the past 11 years, during which over 4,500,000 tons of ore has been milled at this property, the spread between reserve calculations and ore sent to the mill has been slightly over 1 percent in value.

An attempt is made to draw a proportionate amount of ore from each vein coresponding to the ratio of that vein to the total ore reserves.

RATES OF WAGES

The wage rate for an 8-hour day is as follows:

Shift bosses	$7.00
Timbermen	4.80
Helpers	4.24
Scalers	4.80
Machine runners	4.80
Helpers	4.24
Cage and skip tenders	4.80
Pipe and trackmen	4.80
Helpers	4.24
Pumpmen	4.80
Motormen	4.80
Helpers	4.24
Shaftmen	5.52
Blasters	4.80
Diamond-drill runners	5.52
Helpers	4.24
Muckers, nippers, and trammers	4.24
Underground hoistmen	4.80
Samplers	4.80
Helpers	4.24
Steel sharpeners	4.96
Hoisting engineers, first class	5.44

Practically all underground development is let on contract. A fixed price per foot of advance is set. The contractor pays for powder, which is supplied to him with made-up fuse and caps at a fixed price. Timbering is also done on contract, as is all stoping except in special cases.

Measurements are made for contract payments every second week. The men are guaranteed the fixed wage rate, and should any bonus be made the men participate in it on a pro rata basis according to the number of shifts worked. A capable workman is usually sure of earning a substantial increase in pay over the base rates.

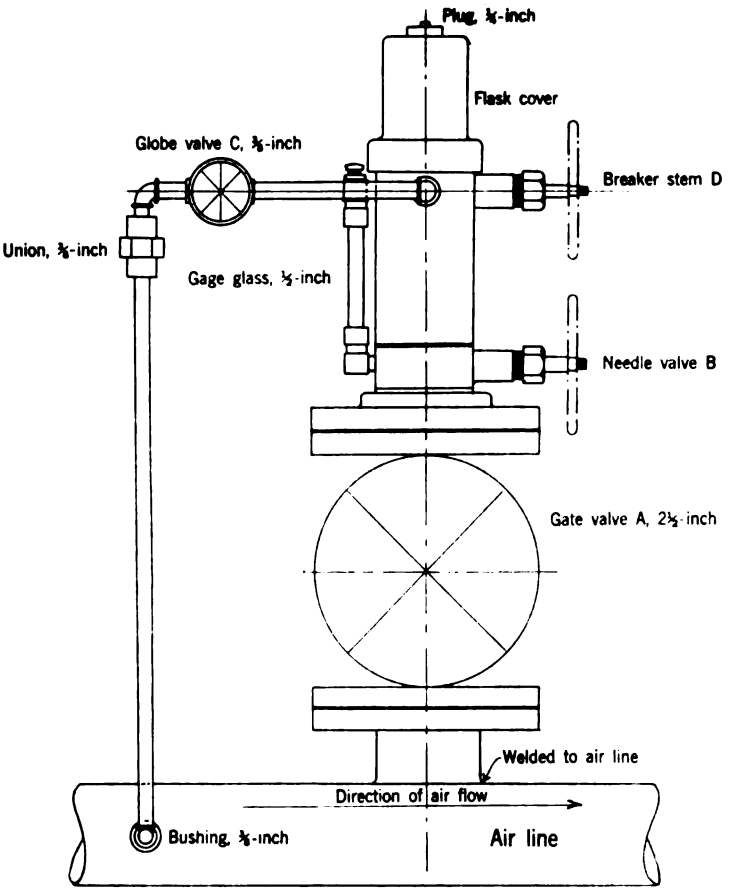
VENTILATION

Ventilation at the mine is natural. Booster fans are used for individual working places

In winter all operating shafts are upcast. To maintain the upward course of the air it has been necessary to admit live steam just below the shaft collar in shafts 5 and 11 in the coldest weather.

In summer all openings are upcast except shaft 11. By means of splits the mine is divided into five separate sections.

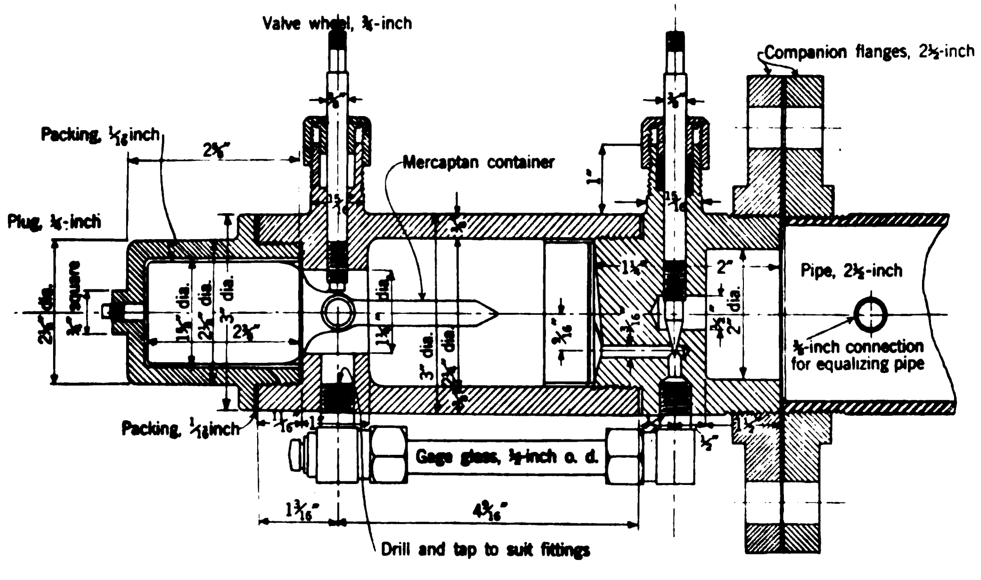
There are drift connections with the older McIntyre workings from shaft 11 on the 1500, 1625, 1750, 1875, 2000, 2125, 2250, 2375 and 2875 levels. Below the 2875 level there is a connection on the 3375 level of shaft 11 workings with the 2875 level of shaft 6.



TO CHARGE
Open valve A; close valve B; close valve C; screw back breaker stem D; remove flask cover, insert vial; replace flask cover and tighten up

TO INJECT
Screw in breaker stem D (to break vial); after liquid appears in glass open wide valve C; open wide valve B; watch glass for discharge of liquid

A



B

Figure 18.—Mercaptan injector: A, general layout; B, detail drawing.

By means of these connections and raises, shafts 4, 5, 6, and 7 provide separate splits for mine ventilation above the 2875 level. Below the 2875 level a 10,000-cubic foot capacity, reversible fan on the 3875 level and an auxiliary fan at the collar of the new internal shaft 12 provide ventilation below the 2875 level, having direct connection to the surface through the 3375 level connection with shaft 6.

Accident-prevention work is done under the direction of a safety inspector, who makes a daily underground inspection. A safety report is made to the general manager each day, and this report must be signed by the heads of all operating departments. Safety meetings are held twice a month at which reports of accidents are read and attention is called to any unsafe practices or conditions in the mine. Two physicians, in the employ of the company, are in attendance twice a day as the men come off or go on shift. A first-aid man is on duty during the night shift.

A safety campaign has been carried on at the mine for several years and has accomplished a definite improvement in the accident rate. Some years ago the rate was one compensation accident of at least 7 days' lost time to 2.7 men per year; in 1927 the rate was one to 3.4; in 1928 one to 5.5; in 1930 one to 15.7 men; and in 1931 one to 15.9 men.

A bonus system was introduced in 1927. A regular shift boss who had no accidents of over 2 days' lost time for a period of 3 months receives $75, a timber boss receives $37, and a track, sample, or diamond-drill boss receives $25 for a similar record. Lost-time accidents which cause a man to lose time in two periods count against the boss for both periods.

The company maintains the following safety and rescue apparatus:

2 H-H inhalators,
12 Burrell gas masks,
12 M.S.A. self-rescuers, and
1 Carbon monoxide detector.

Underground there are -

3 Emergency tool boxes,
1 Set of jacks,
Stretchers,
Blankets, and
First-aid boxes.

There are 100 men in the employ of the company who have taken the first-aid course given under the direction of the St. John's Ambulance Society. Sixty of these men work underground. Two first-aid teams are entered each year in the McCrea Trophy Contest, which is open to the mines of the province; the No. 1 team of 1931 won second place in that year and won the McCrea trophy in 1932.

Six men from the mine are on the Porcupine district mine rescue team. Once a month they practice actual underground work with masks and full rescue equipment.

It is compulsory for the men to wear safety hats, hard-toed shoes and boots, and gloves. This equipment is supplied to them at cost.

There is an installation for injecting ethyl mercaptan in the main air lines at the compressor house. Figure 18,A, is a general layout of the injector installation; Figure 18,B, a detail drawing of the injector.

Special wood-lined powder cars are used for powder distribution. On each level at about 100 feet from the shaft a stub switch is placed, and the powder car remains on the switch until powder distribution is made to the various working places.

In connection with the study of silicosis a dust survey of the mine has been made. Silica content of the country rock ranges from 45 to 65 percent, while vein material varies from sulphides with some silica to practically solid quartz veins. The silica content of the dust of the rocks is relatively lower than in the Rand mines, the results from which are usually made the basis of such comparisons.

The apparatus used to ascertain the silica content of the dusts was the circular konimeter invented by Sir Robert Kotze which is used by the South African Government in making dust surveys on the Transvaal.

The microscope used had a magnification of 160 diameters, and the count was made on dark ground illumination. The safety limit of dust count with dusts from rock of high silica content, such as that of the Rand, has been taken as 300 particles of dust between 0.5 and 10 microns (1 micron equals 1/25000 inch) per cubic centimeter. As Rand ores have probably a 20 to 25 percent higher silica content than McIntyre ores, there is a considerable margin of safety in this figure. Counts were made of dusts in drilling, mucking, tramming, and chute and other blasting operations.

Conditions vary greatly for different parts of the mine, but dust counts on the basis mentioned gave the following general results:

Kind of work	Results	Conditions
Blasting	High	Bad
Drilling	Above limit	Fair
Tramming and chute blasting	Slightly above limit	Good
Mucking	Limit or below	Good
Other work	Below limit	Excellent

To improve dust conditions, the decisions arrived at from the study were that -

(1) Dust production must be prevented.

(2) Dust must be removed from air before it is breathed.

(3) Dust must be diluted by ventilation.

To accomplish these improvements the following measures are used:

(1) All holes are collared wet.
Water connections are used on all pluggers and stopers.
Hitches are cut wet.
All ore and waste passes are bratticed off.
All muck piles are kept wetted down.
Air is not blown in development faces when entering.

(2) Water sprays are used after blasting. (Figure 19 illustrates the connections for water sprays.)
Respirator is used when reloading cut or completing square-up.

(3) Auxiliary ventilation is used at individual faces.

The history of silicotic development at this mine since physical examinations have been compulsory indicates that hazards from dusts may be controlled.

In line with the campaign to combat silicosis the McIntyre, on November 15, 1931, put into commission a mine dry house which has several new features. A tunnel leads from the collar at shaft 11 directly to the dry house, so that the miners in their wet underground clothes are not exposed to the extremes of winter temperatures. In the change house are the mine captain's and shift bosses' offices. Also in the waiting room in front of the shift bosses' office a large safety bulletin board is maintained. On it, in addition to the safety data, are posted items of news and notices concerning athletic events. All employees may be members of the McIntyre Athletic Association on payment of a small fee. The affairs of this association are controlled by a committee on which the employees are represented.

The essential features of the new dry house are the two change rooms, one for the mine clothes and one for the street clothes. In putting off mine clothes for street clothes, or vice versa, a complete change is made. Facilities are provided for shower baths and an exposure to ultra violet rays in a solarium. This latter treatment is equivalent to 3 hours' total body exposure to ordinary sunshine. After changing to their street clothes the men are ready for their walk home in extremes of weather in absolutely dry clothing after the invigoration of a cold shower and sun-lamp treatment.

Lunch boxes are passed from the mine dry house to the street dry house by a belt conveyor through an inspection room. By this method, as every man must pass from one room to the other completely naked and as their lunch boxes are inspected by a company employee, the possibility of high-grading is greatly reduced and the consequent suspicions regarding such practice are eliminated.

To date, while absolute data for comparison are lacking, everything points to the fact that this new dry-house installation is fulfilling the purposes for which it was intended.

PUMPING

The mine is comparatively dry. There is one watercourse which, when tapped by diamond drilling on the 1000 level, gave a pressure indicating a static head from the surface.

Water for drilling purposes is piped from the surface and is from the same system which supplies the towns of Timmins and Schumacher.

All water is drained to shaft 11, and pumps at the 3875, 2875, 1875, and 1000 levels handle the water in a 4-inch column in lifts of approximately 1,000 feet. Triplex 5- by 10-inch pumps are used and are driven by 60-hp. motors.

Summary of costs per ton, 1931

Ore mined and hoisted: 557,104 tons.

	Labor	Supervision	Compressed air	Power	Explosives	Lumber	Other supplies	Total
Development	$0.405	$0.016	$0.021	$0.025	$0.159	-	$0029	$0.655
Mining (stoping)	.891	.038	.031	054	.147	$0.240	.088	1.489
Transportation (underground)	.087	-	-	.005	-	-	.043	.135
General expense (underground)	.357	.036	-	.084	.003	-	.180	.660
Surface (Directly applicable to underground operations)	.104	.004	-	.008	-	-	.018	.134
Totals	1.844	0.094	0.052	0.176	0.309	0.240	0.358	3.073

SUMMARY OF COSTS IN
UNITS OF LABOR, POWER, AND SUPPLIES FOR ONE MONTH[1]

	Tons
Ore hoisted	66,902
Ore from development	5,598
Ore from production	61,304
Total ore broken in development	16,022

	Development	Mining	Total
A. Labor (man-hours per ton):			
Breaking (drilling, blasting, etc.)	1.34	1.52	1.38
Timbering and filling	-	.14	.13
Haulage and hoisting	.45	.33	.31
Supervision	.09	.07	.06
General	.95	.51	.47
	2.83	2.57	2.35
Average tons per man-shift	-	-	3.4
Labor, percentage of total cost	-	-	55.3
Average tons per man-shift:			
Surface labor chargeable to underground operation	-	-	30
B. Power and supplies:			
Explosives (pounds per ton)	3.5	.87	1.09
Total power, horsepower-hours per ton	-	-	25.89
(1) Air compression	-	-	13.12
(2) Hoisting	-	-	10.69
(3) Pumping	-	-	1.43
(4) Ventilation	-	-	.65
Other supplies in percentage of total supplies and power	-	-	49.1
Supplies and power percentage of total cost	-	-	37.9
C. Percentage of total cost	19	81	-

---

[1] Figures based on tons of ore milled (2,000 tons per day at present).

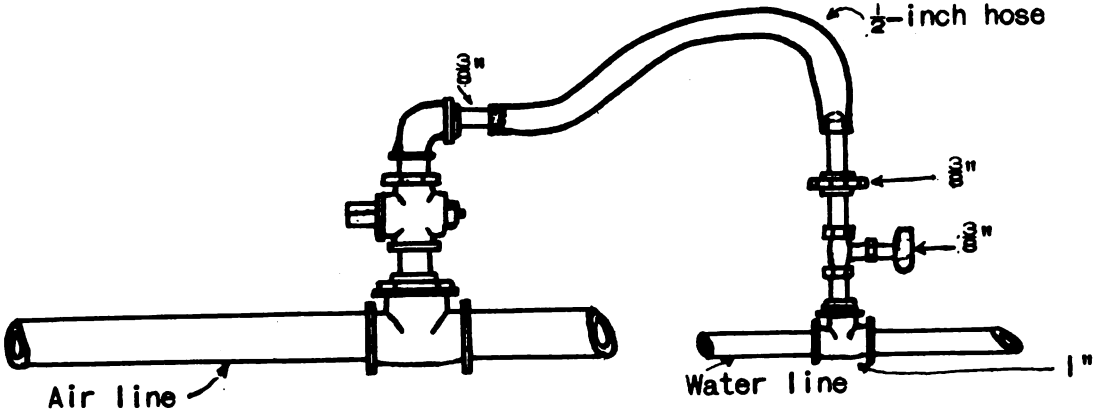


Figure 19.- Connections for water spray